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# MINING METHODS OF KIRKLAND LAKE GOLD-MINING CO. (LTD.), AT KIRKLAND LAKE, ONTARIO.

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# MINING METHODS OF KIRKLAND LAKE GOLD-MINING CO. (LTD.), AT KIRKLAND LAKE, ONTARIO[1]

By J. C. Dumbrille[2]

## INTRODUCTION

The following account of mining practices of the Kirkland Lake Gold Mining Co., (Ltd.), at Kirkland Lake, Ontario, is one of a series of papers being prepared by the United States Bureau of Mines on mining practices, methods, and costs in various districts.

The author has prepared this paper with the help and permission of J. B. Tyrrell and W. M. Sixt, the managing director and the mine manager, respectively.

This mine is operating at a daily rate of about 150 tons.

## HISTORY

The first gold discovery in this camp was in 1911 on a claim which is now part of the Wright-Hargreaves mine. In the following months the Tough-Oakes was staked, and by the summer of 1913 all the present producers were being explored.

In 1915 the Tough-Oakes came into production with a 125-ton mill. Since then the other producers have gradually come into production and have increased until the present output of the camp is approximately $1,700,000 per month.

The mill was built on this property in 1919. Some changes, though no major ones, have been made in the flow sheet in the past year, due to the higher grade of ore being handled.

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6490."

[2] One of the consulting engineers, U. S. Bureau of Mines.

## GEOLOGY

The geology of the camp has been covered in a comprehensive report written for the Ontario Department of Mines by E. W. Todd. The rocks in the camp are all pre-Cambrian, and the overburden is not excessive.

The Keewatin and Temiskaming formations are intruded by Algoman rocks along a major synclinal structure, which is traceable for many miles. The Algoman rocks on this property are lamprophyre, syenite, and porphyry. In the camp there are three Algoman diabase dikes which run transversely to the fault zone and are there displaced. The main mass of rock on this property is lamprophyre. This rock is cut by the more acid syenites and porphyry. These acid types are more predominant at depth.

The faulted zone, which contains the veins of the camp, strikes with the syncline. Movement or displacement on the faulted zone can be measured by observation of the beds of Temiskaming conglomerate and tuff. This displacement is reversed faulting, causing elevation of the south side of the fault. The distance of displacement is approximately 1800 feet vertically and 120 feet horizontally.

Metal-bearing solutions were injected subsequently to this faulting. Postore faulting has not been proved on this property. The deposition of ore apparently depends more upon the character of the fracturing in the rocks than upon any other single factor.

## PHYSICAL CHARACTERISTICS OF ORE AND INCLOSING ROCKS

The fault zone ranges in width from a few feet to 100, and ore may be found on mud slips which occur throughout this zone. In places the ore extends from wall to wall of this zone.

The ore bodies are usually roughly lenticular and longer vertically than horizontally. The ore shoots or lenses in practically all cases rake west going down and occur in echelon formation. They range from 30 to 300 feet in horizontal length and 100 to 500 feet along the long axis. The width varies up to a maximum of 30 feet. Figure 1 illustrates the occurrence of ore shoots within the shear zone.

There is no apparent change in general dip at the present depth of 4300 feet. The majority of the veins worked have been on the hanging wall or footwall of the nearly vertical faulted zone and therefore have a fairly steep dip. Some high-grade veins dip at a much flatter angle.

The ore usually consists of the country rocks impregnated with and replaced by quartz which has been highly fractured.

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The rocks in the ore zone are highly fractured and are cut into blocks by intersecting faults or mud seams. In mining, practically no trouble has been experienced as a result of depth on the vein system itself. However, in crosscutting at the bottom levels it has been found that excessive strain exists in the vicinity of porphyry dikes in the lamprophyre mass. This strain releases itself by rock bursts, which have occasioned some little trouble.

No marked change has been encountered in the character of mineralization, although more tellurides may occur at the bottom levels. The mineralization is very fine, and visible gold rarely occurs.

## METHODS OF PROSPECTING AND EXPLORATION

When the faulted zone is penetrated in crosscutting it is sampled across the full width, and a drift is driven along the most promising fracture. Usually the footwall mud seam is followed, as it has been found that most ore bodies will appear on this fracture at some part of their development. However, there are notable exceptions to this rule.

The face of the drift is sampled daily. Daily test holes are drilled into the walls so that development may be directed most intelligently. After drifting is completed on a level, diamond drilling is done laterally to explore the full width of the ore zone. Any favorable core samples are checked up by crosscutting to them.

In addition to diamond drilling it is the usual practice to run crosscuts through the fractured zone at intervals of approximately 300 feet.

## METHODS OF SAMPLING AND ESTIMATION OF TONNAGE AND VALUES

The test-hole samples which are taken in exploration drifts are supplemented by test-holing in stopes. All cut samples taken underground are cut with the moil. The ore usually can be determined only by assaying, and therefore all samples are in short sections.

Level plans are made on a scale of 1 inch to 30 feet. The geological data and drift assays are plotted on tracings made from these plans. Vertical projections of the stopes are made on a scale of 1 inch to 10 feet, and stope assays are plotted on them. These stope plans are made from measurements taken by the sampler; and stopes are surveyed with the transit only in special cases.

Ore and rock are estimated at 12 cubic feet per ton in place and 20 cubic feet per ton broken.

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## METHODS OF DEVELOPMENT AND MINING

### Exploration

During the past year a very heavy development program has been carried on. A system of raises has been driven, to complete an outlet independent of the shaft, and this work has been charged to development and exploration. Upon adding this to an unusual amount of shaft sinking and other development, the total is very high. The average number of machines per shift has been 7, and of these 4 and often 5 have been on development work.

The early development of the mine was from No. 1 shaft, which now extends to the 900-foot level. Later the present main shaft was sunk to the 2,475-foot level. A winze was sunk from a point some 400 feet south of the main shaft on the 2,475-foot level. This winze is now down to the 4,300-foot level and is being sunk deeper. All shafts and winzes are vertical. The level interval is 125 feet. Figure 1 is a sketch map of the 3,875-foot level, showing the method of development and the shape and occurrence of the ore bodie

### Shafts and Winzes

No 1 shaft is a 2-compartment shaft, 9 feet 8 inches by 5 feet 8 inches, outside dimensions. This shaft is now used only as a downcast air passage and emergency ladder road. The main shaft is a 3-compartment shaft and is practically the same in detail as the winze. The winze from the 2,475-foot level is 3-compartment. The outside dimensions are 14 feet 4 inches by 6 feet 4 inches. The wall plates and end plates are of 8 by 8 inch and the dividers 6 by 6 inch spruce timbers. The sets are normally placed on 6-foot centers, and $\frac{3}{4}$-inch hanging rods are used. Bearing timbers 10 by 12 inches are placed below each level.

In the upper parts of the mine it was not found necessary to lag the shaft, but at the lower depths the shaft is lagged completely and filled with cedar blocking. It has been found necessary to shorten the spacing of the sets to 4-foot, 3-inch centers, due to the fact that pressure bursts occur at or near the contacts of porphyry dykes in the lamprophyre, and it is found necessary to install a set of timbers as soon as there is headroom in the bottom of the winze. A steel blasting set of 16-inch I beams is hung below the timber, and a bulkhead is maintained over the men one set above the blasting set.

The round, as drilled in the hard rock at the present horizon, consists of 50 holes. The cut is a V in the center of the shaft. The cut is drilled with 8-foot steel and the square-up with 6-foot.

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## CORRECTED FIGURE 1

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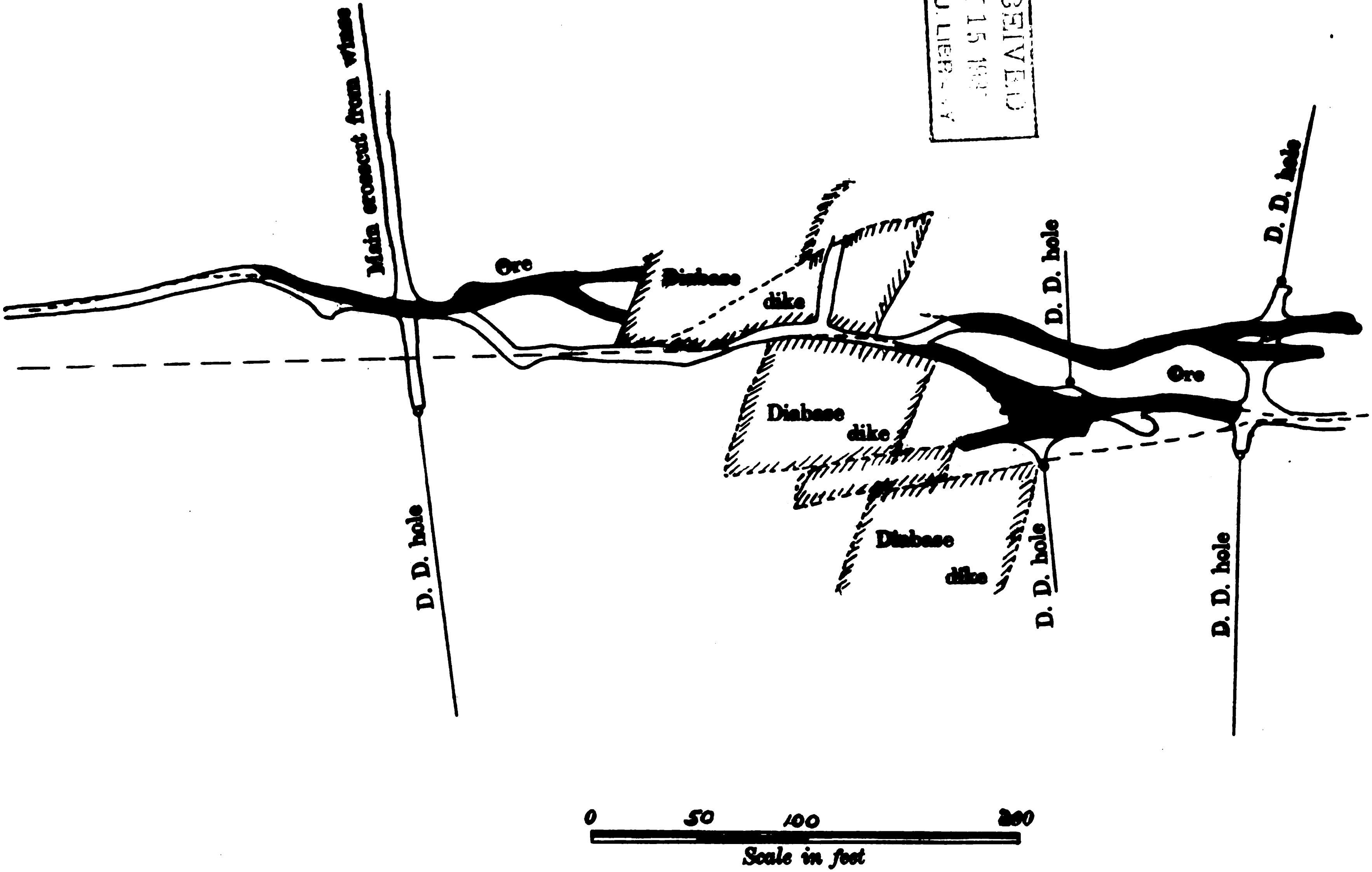

Figure 1.—Sketch map of 3,875-foot level, showing method of development

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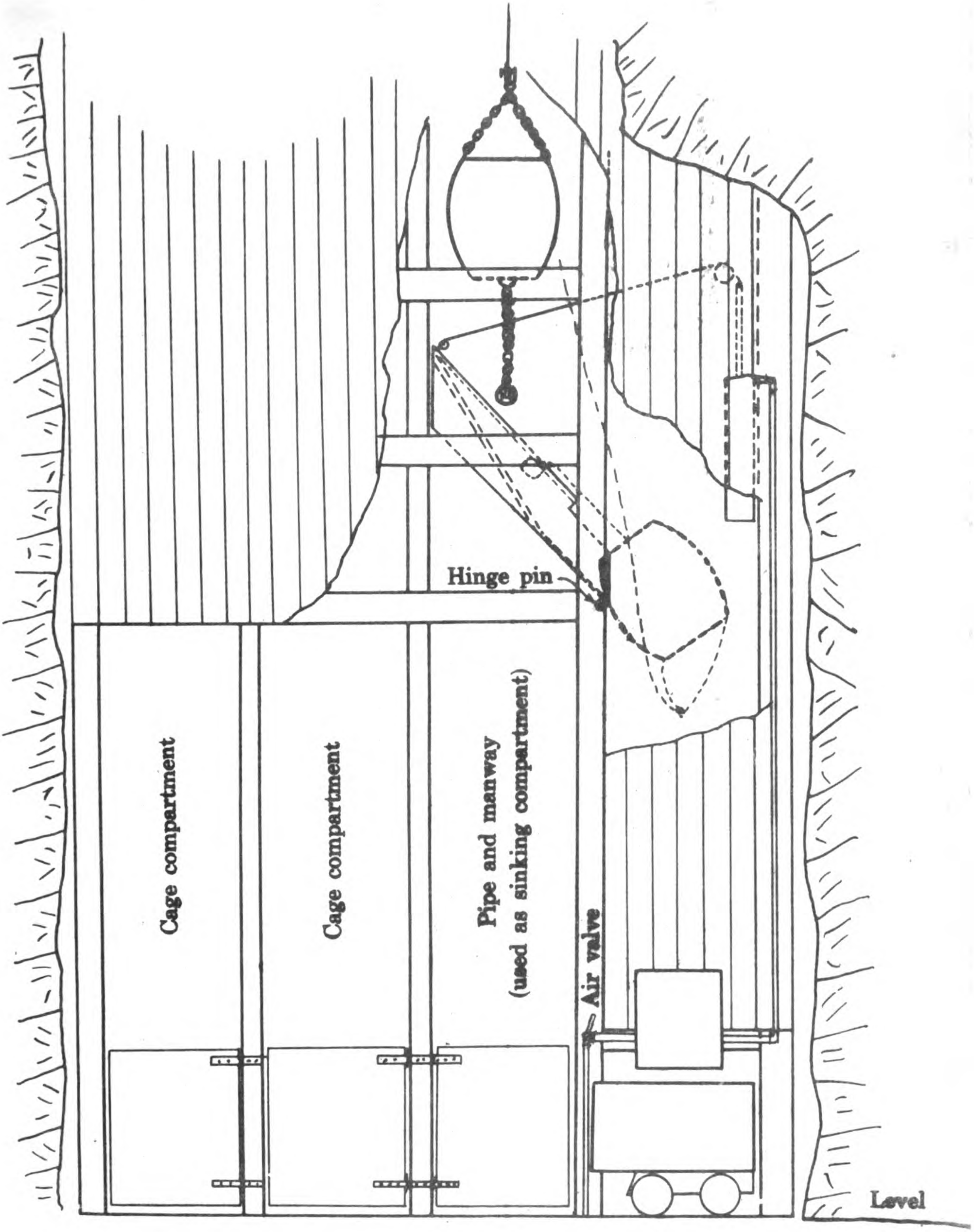

**Figure 2.– Sketch of dumping arrangement for sinking**

The machines used are heavy drifters, S70 Rand (4-inch piston) and SL15 Holman ($3\frac{3}{4}$-inch piston). It has been found that lighter machines can not compete with these in the hard rock. Two machines are used on crossbars. The steel is 1 1/8-inch hexagon. Five and six changes are employed, with a gage interval of 1/8 inch. Smooth shanks and tappets are used.

The work has been carried on recently in three 8-hour shifts a day. Each shift does whatever work is at hand. Delays due to the uncertain nature of the ground have been found to upset any schedule of operations which might be planned. Each shift consists of a leader and four men. Drilling off takes approximately 9 hours.

Blasting is done with electric delay caps from a 110-volt A.C. circuit. About 150 pounds of 40-per cent Forcite (gelatin dynamite) is used to a round. Mucking out a round requires about 12 hours.

Muching is done by hand into 15-cubic foot buckets. No crosshead is used in hoisting. The compartment used for hoisting is lined flush with 1-inch plank, and a $\frac{3}{4}$-inch nonrotating cable is used in hoisting. The bucket has a chain bail and is so shaped as to offer the least chance of catching in the timbers. Dumping is accomplished by means of a ball and chain attached to the bottom of the bucket. A door is lowered across the compartment by means of a windlass or air piston, and the bucket is lowered until the ball is engaged by a slot in the door. On further lowering, the bucket drops into the bin and dumps. A sketch of the dumping arrangement is shown in Figure 2.

Timbering, which must be done after every round, requires about 8 hours. In the main shaft the cycle was much shorter. A round a day could be taken out on two shifts, and timbering was done once a week, usually on a special Sunday shift.

Station cutting at the lower levels is simple. The station is driven 7 feet high, and the back scales off sufficiently to give headroom at the shaft. After sloughing off the station back is timbered.

## Drifts and Crosscuts

Crosscuts are driven 7 feet high by 6 feet wide. No timbering is necessary, as the ground will stand. At lower levels frequent scaling is imperative until the roof becomes arched.

Where speed is essential the crosscuts are driven on a 3-shift program. Four men on a shift - two machine men and two muckers - drill, blast, and muck one round per 8-hour shift. The men either use a crossbar or muck back and set up a vertical bar, at their own choice. The heavy-type machines employed in the shaft must be used to gain the drilling speed necessary in this work.

Where speed is not so essential a continuous 3-shift program is adopted. There are two machine men on each shift, and they drill or muck as required. More advance per round can be made this way, as the drilling shift does not need to muck back and can place its holes to more advantage. About 75 pounds of 40-per cent Forcite is used per round.

Drifting is done in the same manner as crosscutting, although the width of drift may vary according to the width of ore. The drilling is usually easier, and fewer holes are required to break the same amount of ground. The advance per shift is greater than in crosscutting.

## Raising

Raising is done with drifters of the lighter type. Practically all raises are inclined at 45 or 50° and are untimbered. The dimensions are about 5 feet by 5 feet. A diamond cut is drilled.

## Stoping

The stoping carried on in the past has been all shrinkage stoping with timbered backs. The mine is experimenting with other methods at present. Some method of cut and fill will probably be adopted eventually. In one small stope a modified cut-and-fill method with hand sorting of ore is being used. This is a special method adopted to fit special conditions.

In shrinkage stoping the backs of the drifts are all timbered, and chutes are spaced on 10-foot intervals. Round spruce obtained locally is used in stope timbering. Manways, 4 by 4 feet, are carried up to 50 feet above the level. Raises are driven in the ore from this height to the level above. Work is then carried on from the upper level, and one manway kept open for the disposal of steel and for other service.

In stoping it is the practice to leave about 2 feet of ore on the walls. This will slough off and keep the walls in better shape than if blasted off. The walls are usually faulted and fairly smooth, but if they are broken the rock will not stay in place, as it is very blocky.

Drilling in stopes is done with drifter machines having 3¼ inch pistons. These machines are somewhat lighter than those employed in development. Flat breast holes are drilled. The depth of holes ranges to 10 feet, depending upon the width of the ore body.

Blasting is done at the end of the shift with 40-per cent Forcite, No. 8 detonators, and safety fuse.

Except in the instance mentioned no sorting is done in the stopes, and blockholing is unnecessary.

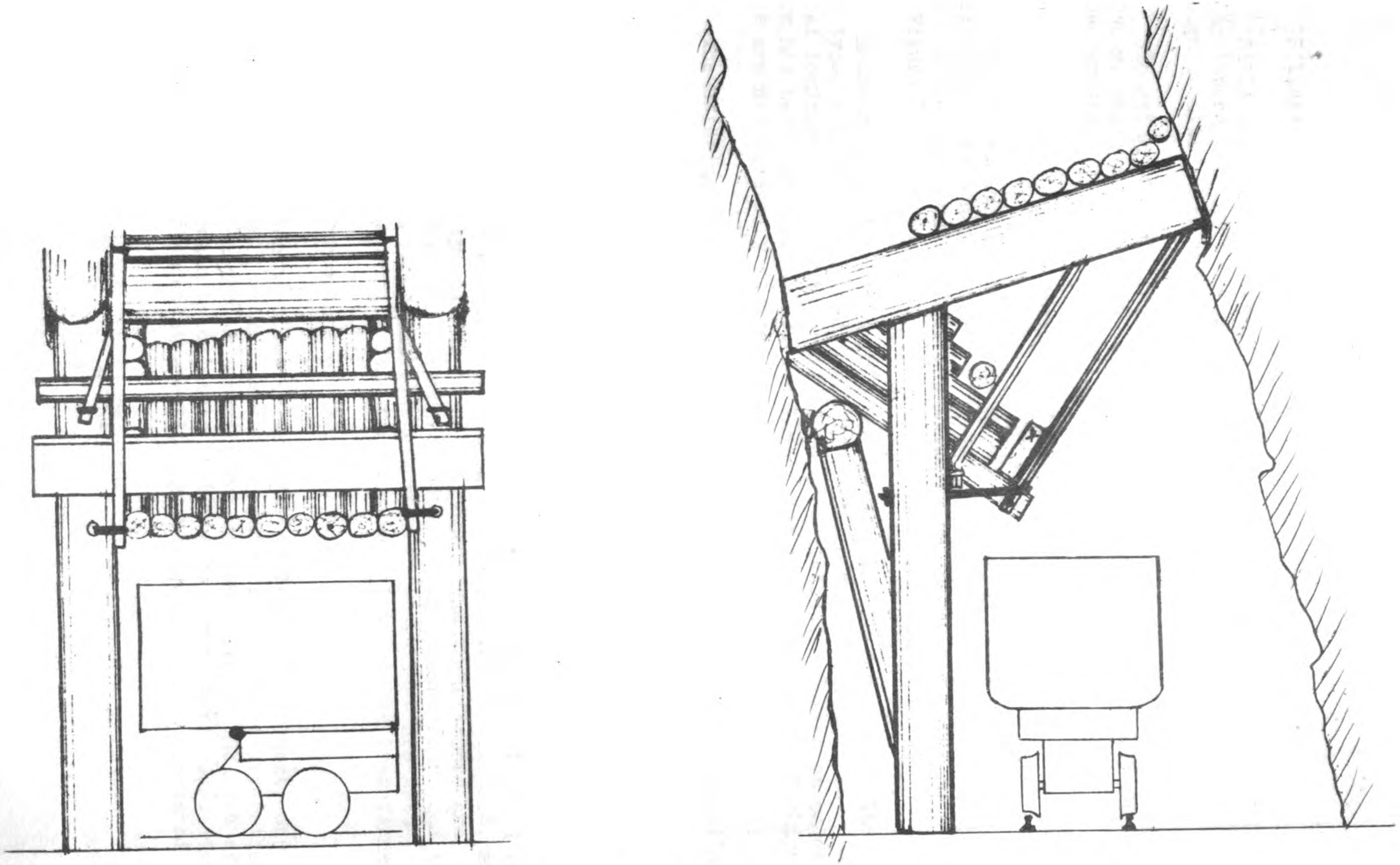

Figure 3.– Standard ore chute

Pillars are left very rarely.

Sills are recovered by mining directly below the old timber and retreating toward the shaft. The empty stope above will usually stand up long enough to allow this procedure.

Sloughing of waste from the walls is mostly in the form of very large blocks, which will hang in the stope and allow practically all the ore to be removed.

## Underground Haulage

The chutes in the stopes are of simple, cheap construction. Spruce lagging, 6-inch diameter, is used for the bottom and sides and the headblock is 12-inch round. These are illustrated in the accompanying sketch, Figure 3.

Hand-tramming is used throughout the mine. The cars are of the end-dump type, roller bearing equipped, of 20-cubic feet capacity. No mechanical loading equipment is used. Cars are hoisted to surface on cages. Provision has been made in the station layout for the installation of skips, but these are not necessary at present.

The hoisting is in two stages, as previously noted. The installation on the main shaft is an Ingersoll-Rand hoist of the motor-driven double-drum type. The rope speed is 1000 feet per minute. The hoist capacity is 12,000-pounds pull. Langlay flattened-strand cable 1 inch in diameter is used. Two cages run in balance. One drum is clutched to permit single hoisting if necessary.

The winze hoist is of the same type, with a rope speed of 1500 feet per minute and a capacity of 12,000 pounds. Both drums are clutched in this hoist. The roof of the hoist station on the 2475 level is timbered, and this timber is supported by columns of 6-inch pipe. The transformers are housed in a special chamber with a fireproof door. This transformer room is ventilated by a small fan which supplies air through pipes that are automatically sealed if an abnormally high temperature exists at the transformers.

## Percentage of Extraction

In shrinkage stoping dilution is high, at least 30-per cent, due to the character of the wall rock. As the majority of the stopes are rather small it will not pay to change at present, but in large stopes the loss of ore and the lowering in grade by dilution would be so great as to necessitate use of another system. At present practically all ore broken is recovered.

## WAGE, CONTRACT, AND BONUS SYSTEMS

No contract or bonus system is employed in this mine, as a regular practice. In special instances a footage bonus is paid in development work. All men at the same type of work receive the same pay, regardless of length of service.

The scale of wages follows:

| | |
|---|---|
| Surface.................. | $3.75 |
| Helpers and muckers ..... | 4.25 |
| Machine men, timber men.. | 4.75 |
| Shaft men ............... | 7.00 |

## VENTILATION

Natural ventilation is supplemented by a fan on the 3,225-foot level. During the past year a series of ventilation raises was driven to form a downcast air passage. These raises connect the old No. 1 shaft with the fan. Where the downcast passage is connected with the upcast side by old workings a door is installed. Thus both the main shaft and winze are upcast. The fan forces the fresh air through a 12-inch galvanized pipe to the bottom of the winze. A connection is available at each intermediate level to allow for taking off a regulated amount of downcast air. The fan capacity is 3500 cubic feet per minute.

Efficient ventilation is necessary at depth due to the higher temperature of the rock. This increase in temperature, combined with the increased barometric pressure, causes uncomfortable working conditions if a supply of fresh, cool air is not available. The increase in rock temperature in this mine is 1° for every 148 feet of depth. The rock temperature at 4300 feet is 69° F.

Provision is made for a doubling up of the air piping in the shaft and an increase in capacity of the fan.

## FIRE HAZARD

Due to mine conditions the fire hazard underground is small. All waste timber and powder-box refuse are brought to surface.

No magazine is maintained underground, and no more than 300 pounds of explosive is stored in any one place. The amount of explosives and detonators required is taken underground daily.

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The winze hoist station, which is timbered, is protected from fire by a sprinkler system which is turned on daily. This also wets down the upper timbers in the winze. Both main shaft and winze are dripping wet. This is advisable not only for fire protection but to prevent decay in the timbers, as the warm upcast air would destroy dry timber in a short time.

The main fire danger is that men would be trapped underground by a surface or brush fire. A sprinkler system is installed in the main-shaft headframe, and extensive fire-fighting equipment is maintained on the surface.

Refuge stations are installed at intervals throughout the mine. These consist of abandoned drifts and workings, walled off and made gas-proof by doors. Food and water are kept in these stations at all times, and the men are trained to go there at the alarm of fire rather than attempt to climb to the surface.

The alarm can be given in the mine by the introduction of Ethyl mercaptan, a stench compound, into the compressed-air stream at the surface.

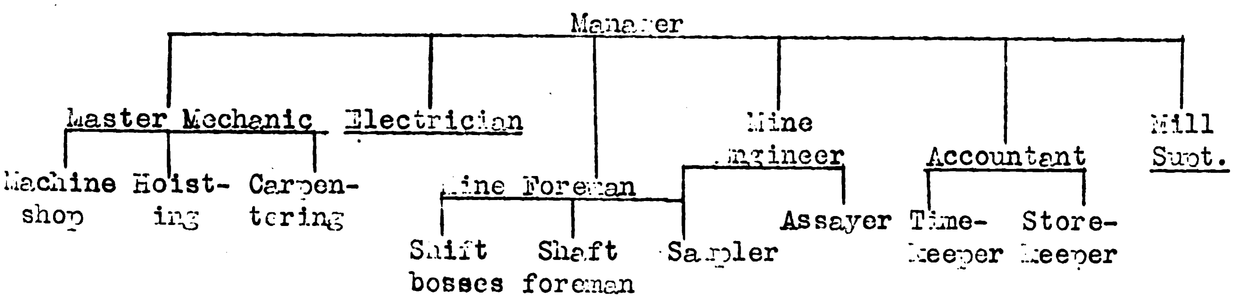
## SAFETY METHODS AND FIRST AID

All foremen and bosses are held responsible for the safety of the men under their charge.

A first-aid room is maintained adjoining the mine office, and certain members of the office staff are trained in first aid.

## ADMINISTRATION ORGANIZATION

The mine organization is as follows:



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The manager has direct contact with all department heads.

The master mechanic attends to all mechanical repair work and maintenance. He is machine-shop foreman and supervises hoisting, general surface work, and carpentering.

The electrician is responsible for all electrical work.

The mine foreman is directly responsible for all mining details. All men going underground are under his direction. He has a shift boss on each shift and a shaft-sinking foreman to assist him.

The mine engineer oversees all engineering work. Surveying, assaying, direction of diamond drilling, and keeping of plans and assay records are in his department. The sampler works under the supervision of the engineer and mine foreman jointly.

The mill superintendent has control of crushing, milling, and refining.

The mine accountant has a timekeeper and storekeeper under his supervision.

## COSTS

In Tables 1 and 2 the costs are based upon a full month's work,—mining, raising, sinking, crosscutting, and drifting. In Table 3 representative periods of development work were taken from the same month.

Table 1.- Summary of Costs

Kirkland Lake gold mine. Period, June 1 to July 1, 1930.

Hoisted during period - 4,108 tons, ore.
do 6,531 tons, ore and waste.
Broken during period - 4,712 tons, ore.
Mining method - Shrinkage stoping

Underground costs

| | Per ton of ore hoisted | | | | | | | | Per ton of ore and waste hoisted |
|---|---|---|---|---|---|---|---|---|---|
| | Labor | Supervision | Comp. air, drills, & steel | Power | Explosives | Timber | Other supplies | Total | |
| Development | $1.49 | $0.10 | $0.38 | - - | $0.39 | $0.42 | $0.09 | $2.87 | $1.78 |
| Mining | .77 | .07 | .23 | - - | .46 | .03 | .06 | 1.62 | 1.02 |
| Transportation and hoisting | 1.17 | .06 | - - | $0.34 | - - | - - | .04 | 1.61 | 1.01 |
| General: | | | | | | | | | |
| (a) Underground | .04 | .01 | - - | .05 | - - | - - | .04 | .14 | .09 |
| (b) Surface | .45 | - - | - - | - - | - - | - - | .07 | .52 | .33 |
| Total | 3.92 | .24 | .61 | .39 | .35 | .45 | .30 | 6.76 | 4.25 |

Table 2.- Summary of Cost in Units of Labor, Power, and Supplies

Kirkland Lake gold mine. Period, June 1 to July 1, 1930.

Hoisted during period - 4,108 tons, ore.
do 6,531 tons, ore and waste.

Mining method - Shrinkage stoping

| | Development | Mining | Total |
|---|---|---|---|
| A. Labor: | | | |
| Breaking ..... man-hours per ton of ore hoisted | 1.16 | 0.90 | 2.06 |
| Timbering .... do | .46 | .07 | .53 |
| Shoveling .... do | 1.08 | .48 | 1.56 |
| Hauling ...... do | .57 | 1.01 | 1.58 |
| Supervision .. do | .10 | .13 | .23 |
| General ...... do | .65 | .22 | .87 |
| Total labor. do | 4.02 | 2.81 | 6.83 |
| Average per man shift........................tons | 1.99 | 2.85 | 1.17 |
| Labor,...................per cent of total cost | 60.00 | 65.00 | 62.00 |
| B. Power and supplies: | | | |
| Explosives, 40% - | | | |
| Forcite.........lbs. (per ton of ore hoisted) | 2.18 | 2.54 | 4.72 |
| Timber.......board ft. ( do ) | 2.48 | 1.75 | 4.23 |
| Total power.....hp. h. ( do ) | 61.8 | 50.00 | 112.8 |
| Other supplies...per cent of total supplies and power | 23.9 | 20.2 | 44.1 |
| Supplies and power.......per cent of total cost | 40 | 35 | 38 |
| C. Per cent of total cost | 58.6 | 41.4 | 100 |

Table 3.- Summary of Development Costs in Units of Labor, Power, and Supplies

Kirkland Lake gold mine. 1930.

| | Drifting | Sinking | Raising |
|---|---|---|---|
| Size of excavation ........................ | 6' x 7' | 6'4"x14'4" | 5' x 6' |
| Timbered, or not ........................ | Not | Timbered | Not |
| Physical properties of rock ............... | Hard to drill, easily broken | Hard to drill and break | Hard to drill and break |
| A. Labor: | | | |
| Breaking ............ man-hours per foot | 4.0 | 12.7 | 12.1 |
| Timbering ........... do | | 11.2 | .2 |
| Shoveling ........... do | 3.9 | 16.9 | |
| Hoisting ............ do | 2.6 | 28.8 | 3.8 |
| Supervision ......... do | .5 | 1.4 | 1.1 |
| General ............. do | 1.0 | 12.8 | 1.7 |
| Total labor ....... do | 12.0 | 83.4 | 18.9 |
| Feet per 8-hour shift .................. | 5.2 | 1.0 | 1.3 |
| Feet per 8-hour man-shift .............. | .666 | .096 | .423 |
| B. Power and supplies: | | | |
| Explosives, 40% Forcite.....lbs. (per ft. advanced) | 16.9 | 31.7 | 20.0 |
| Timber......board ft. ( do ) | | 276.0 | |
| Power - air compressor.......hp. h. ( do ) | 129 | 376 | 159 |
| Hoisting ...... do ( do ) | 76 | 852 | 101 |
| Other supplies, per cent of total power and supplies ........................ | 38.3 | 22.3 | 25.2 |
| C. Labor................. per cent total cost | 50.6 | 63.6 | 64.0 |
| Power and supplies... do | 49.4 | 36.4 | 36.0 |

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